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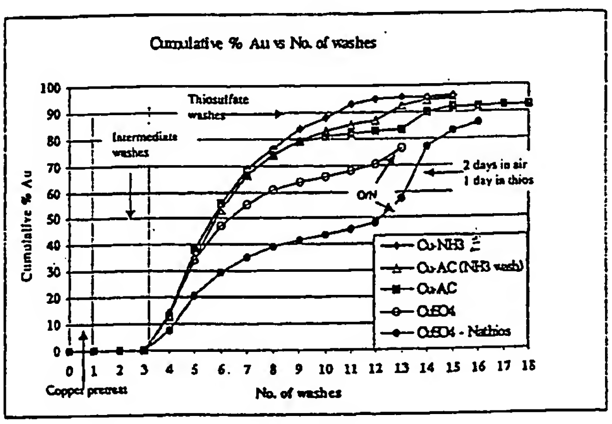


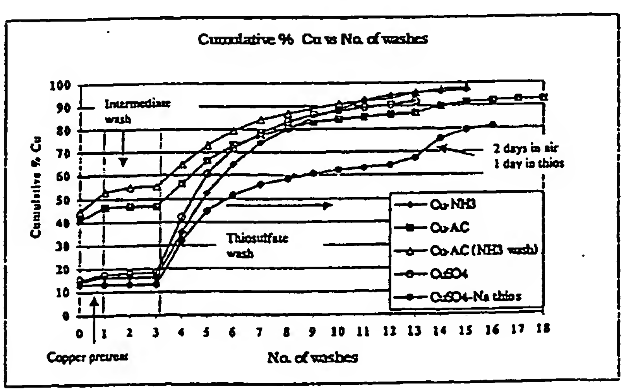


(57) Abstract: A process for leaching precious metals from material containing precious metals, such as oxidic and sulfidic gold-bearing ores, is disclosed. The process includes the steps of: (i) leaching the precious metal with a leach solution containing a thiosulfate-based lixiviant; (ii) treating the material by oxidising precious metal in the material into a form that is leachable in a subsequent leaching step; and thereafter as a separate step.

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# Recovery of Precious Metals

The present invention relates to thiosulfate leaching of material containing precious metals.

The present invention relates particularly to thiosulfate leaching of gold from gold-bearing material, such as ores and concentrates of ores.

It is known to extract gold from ores using thiosulfate-based lixivient systems. US patents 4,369,061 and 4,269,622 to Kerley describe processes which include lixiviating with an ammonium thiosulfate leach solution containing copper to recover gold from ores, particularly difficult-to-treat ores containing copper, arsenic, antimony, selenium, tellurium and/or manganese. US 4,654,078 to Perez et al discloses a modification of the process disclosed in US patent 4,269,622 and is based on lixiviating ores with copper-ammonium thiosulfate in a solution that is maintained at a minimum pH of 9.5. Other known processes that are based on the use of thiosulfate lixiviants include US patent 5,785,736 to Thomas et al (assigned to Barrick Gold Corporation) and US patent 5,354,359 to Wan et al (assigned to Newmont Gold Co).

An object of the present invention is to provide an alternative process for leaching precious metals, such as gold, using thiosulfate-based lixiviants.

According to the present invention there is provided a process for leaching precious metals from material containing precious metals, which process includes the steps of:

(i) treating the material by oxidising precious metal in the material into a form that is leachable in a subsequent

leaching step; and thereafter as a separate step

(ii) leaching the precious metal with a leach solution containing a thiosulfate-based lixiviant.

The present invention is based on the realisation that high levels of precious metal recovery can be achieved on a cost-effective basis by carrying out precious metal oxidation and precious metal leaching as separate steps.

The material may be any material that contains precious metals.

The present invention relates particularly to materials in the form of ores and concentrates of the ores.

Preferably, the ores and concentrates are gold-bearing ores and concentrates. The gold may be contained in oxidic or sulfidic ores.

In one embodiment treatment step (i) includes forming agglomerates of the precious metal-bearing material and an oxidant.

Preferably the agglomerates are formed by contacting the material and a solution containing the oxidant.

More preferably this embodiment includes forming agglomerates of the material, a binder, and the oxidant.

More preferably the agglomerates are formed by mixing the material (such as an ore or concentrate of the ore) and the binder and thereafter contacting the mixture with a solution containing the oxidant.

Preferably, this embodiment includes curing the agglomerates.

Preferably the curing step is carried out in air for a period of at least 24 hours.

The treatment step (i) may include forming agglomerates of the precious metal-bearing material and the oxidant and a thiosulfate-based lixiviant.

In another embodiment the treatment step (i) includes forming agglomerates of the precious metal-bearing material (with or without a binder) and thereafter contacting the agglomerates with a solution containing the oxidant.

The treatment step (i) may include contacting the agglomerates with a solution containing a thiosulfate-based lixiviant.

In a further embodiment the treatment step (i) includes contacting the material (without agglomerating the material first) with a solution containing the oxidant.

The treatment step (i) may include contacting the material with a solution containing thiosulfate-based lixiviant.

In each of the above embodiments, preferably the amount of the solution of the oxidant is relatively small, typically between 10 and 20%, more preferably, between 12 and 15%, by weight of the weight of the precious metal-bearing material.

In each of the above embodiments, the treatment step (i) may include treating the material with ammonia or

an ammonium salt, such as ammonium carbonate, to stabilise the oxidant.

The oxidant may be any soluble source of copper ions.

Preferably, the oxidant is selected from the group consisting of copper sulfate, copper salt, and ammonium complex of divalent copper.

The thiosulfate lixiviant may be any suitable soluble thiosulfate compound.

Preferably the thiosulfate lixiviant is selected from the group consisting of sodium thiosulfate and ammonium thiosulfate.

The binder may be any suitable binder, such as a cement or an organic binder.

The process of the present invention may be carried out under any suitable pH conditions. In this connection, the applicant has found in experimental work that the subject process can be operated over a wider pH range than prior art processes. Moreover, the applicant has found that the subject process is more flexible with operating pH than a number of prior art processes and consequently pH adjustment may not be necessary - as is the case in these prior art processes.

The present invention may be carried out on a heap of precious metal-bearing material, such as gold-bearing ores and concentrates of the ore, by:

(i) passing the solution of the oxidant through the heap;

(ii) allowing the oxidant solution to drain from the heap;

(iii) passing the leach solution containing the thiosulfate-based lixiviant through the heap; and

(iv) allowing the leach solution containing leached precious metal to drain from the heap.

The above sequence of process steps may be repeated as required to maximise recovery of precious metal from the heap.

The process may include a further step of processing the oxidant solution that drains from the heap to recover the oxidant.

Preferably this step further includes recycling the oxidant to the process.

The process may also include a further step of treating the precious metal-bearing leach solution that drains from the heap to recover precious metal, such as gold, from the solution.

Preferably, this step also includes recycling thiosulfate-based lixiviant to the process.

The present invention is not confined to process precious metal-bearing material in a heap and, by way of example, extends to other processing options such as continuously stirred tank reactors.

The process of the present invention can be applied to both oxidic and sulfidic ores.

In the case of sulfidic ores, the conventional wisdom in the industry is that such ores are refractory and that the sulfidic content of the ores must be at least partially oxidised. However, it has been surprisingly found by the applicant that the process of the present invention can be used to selectively oxidise the precious metal in the ore while minimising or substantially avoiding oxidation of the sulphide ore to sulfate.

The applicant has carried out experiment work on gold-bearing oxidic and sulphidic ores. This experimental work is discussed below.

The experimental work included the following basic process steps:

Step 1 Copper pretreatment

A solution containing cupric ion (either as copper, copper diammine or copper tetrammine) in a predetermined concentration was prepared by dissolving a predetermined weight of anhydrous copper sulfate in a known amount of water. To this solution was added either ammonia (so as to form copper tetrammine) or ammonium carbonate (AC) or bicarbonate (ABC) (so as to form copper diammine). This cupric solution thus prepared was contacted with the ore for a fixed period before separation by filtration (small scale) or natural draining (columns).

Step 2 Intermediate wash (Optional)

If an intermediate wash was used, a predetermined volume of a wash solution (either water or ammonia -0.87M) was contacted with the filtered/drained ore for a fixed period before further filtration/draining.

Step 3 Thiosulfate wash

The copper pretreated and (when performed) washed ore was then contacted with a predetermined volume and concentration of either ammonium or sodium thiosulfate solution for a fixed period before filtration or draining. Thiosulfate washing was repeated until little or no Au was detected in the collected filtrate. In some instances the ore was left in for extended periods between washes.

EXAMPLE 1.

This example relates to small-scale leaching of high-grade oxide ore (~ 250ppm Au)

The objective of this experimental work was to investigate at ambient temperature the influence of:

(i) using $CuSO_4$ as a source of $Cu^{2+}$ as opposed to different ammine systems ($Cu-NH_3$ to yield $Cu(NH_3)_4^{2+}$ or Cu-AC to yield $Cu(NH_3)_2^{2+}$ );

(ii) using sodium thiosulfate rather than ammonium thiosulfate; and

(iii) exposure to air between sequential thiosulfate washes.

Table 1.1 summarises the series of experiments performed.

Table 1.1

| Copper Pretreat | | Intermediate wash | Thiosulfate wash |
|---|---|---|---|
| Pretreat | Copper species | | |
| Compare copper species | | | |

| | | | |
|---|---|---|---|
| $Cu\text{-}NH_3$ | $Cu(NH_3)_4^{2+}$ | Water | ammonium thiosulfate |
| Cu-AC | $Cu(NH_3)_2^{2+}$ | Water | ammonium thiosulfate |
| Cu-AC | $Cu(NH_3)_2^{2+}$ | Ammonia | ammonium thiosulfate |
| $CuSO_4$ | $Cu^{2+}$ | Water | ammonium thiosulfate |
| Compare thiosulfate type | | | |
| $CuSO_4$ | $Cu^{2+}$ | Water | sodium thiosulfate |

The following is a summary of the experimental conditions.

(i) Wt of ore used (g, dry basis): 64

(ii) Copper pretreatment

- wt. of copper sulfate (g):1.0(0.025M)
- Total pretreat volume (ml): 250
- Contact time with ore before filtration (min):15
- No. of washes: 1

(iii) Intermediate Wash(when used)

Water:

- Total Volume (ml): 300
- Volume per wash (ml):100
- No of washes: 3

Ammonia solution:

- Total ammonia pretreat volume (ml): 250
- Concentration (M): 0.87
- No. of washes: 1

(iv) Thiosulfate wash

- Volume per wash (ml):100
- wt of ammonium thiosulfate(s)(g/100 ml wash)(when used): 3.7 (0.1M)
- wt of sodium thiosulfate pentahydrate(s)(g/100 ml wash)(when used): 6.2 (0.1M)

- Contact time of wash soln. with ore before filtration (min):5
- No of washes : determined by Au content in filtrate (usually ~ 8 to 10)

Results are presented in Figures 1.1 and 1.2. These Figures are plots of cumulative %Au or Cu recovered in solution versus the number of washes respectively. Where modifications to the usual sequence in sequential leaching occurred these are highlighted in Figures 1.1 and 1.2.

Conclusion

- In all cases with Cu pretreatment (of any form), the overall Au extraction level is either approaching or exceeding 90%. This suggests that high extraction levels may be achieved with the process of the present invention regardless of the form of the cupric ion.

- The rate and extent to which copper desorbs mimics the trends apparent in gold extraction.

EXAMPLE 2.

This example relates to leaching of as received and agglomerated low-grade oxide ore (~ 6ppm Au) using columns.

The most likely field application of the process of the present invention for low to moderate-grade ores would be as a heap or vat leach.

In order to investigate this process application, a series of columns were fabricated using PVC tubing (D=50mm, L = 350-400mm) and packed with 1 kg of ore (dry weight basis) as illustrated in Figure 2.1. Column leaching (which is a form of heap leaching) was then performed using

the process of the present invention and, to assess its applicability in the field, several trials of varying chemical configuration were performed.

In general, columns were filled (to completely cover the bed) by pumping (from the bottom) or spraying (from the top) a predetermined volume of liquid (either pretreatment or leach). After soaking (usually between ~ 8 and 24h), the liquid was allowed to drain and the ore rested (usually between 1-3 days) before the next soak and rest cycle was begun. Washings were collected and analysed for Au and Cu by AAS.

The column leach trials involved the use of two ore forms, generally referred to as:

(i) the as-received ore; and

(ii) agglomerated ore, where the ore was agglomerated with cement only (usually using 5-6 kg of cement/t of ore.)

To determine the efficiency of column leaching using the process of the present invention (without the intermediate washing step) of a low grade oxide (~ 6ppm Au) ore by varying:

(i) the form of the ore : agglomerated vs as-received (non-agglomerated);

(ii) the form of copper in pretreatment: copper tetrammine vs copper sulfate; and

(iii) the amount of copper in the copper pretreatment step.

The following table (Table 2.1) summarises the experimental matrix performed.

Table 2.1

| Column No. | Ore type | Weight (kg) (dry basis) | Copper Pretreatment Form of $Cu^{2+}$/ concentration equivalent of $CuSO_4$ in g/l) | Thiosulfate Leach Concentration (M) |
|---|---|---|---|---|
| Compare the form of copper in pretreatment | | | | |
| C1 | Agglomerated | 1 | Tetrammine (4g/l) | 0.1 |
| C2 | as received | 1 | Tetrammine (4g/l) | 0.1 |
| C3 | Agglomerated | 1 | $CuSO_4$ (4g/l) | 0.1 |
| C4 | as received | 1 | $CuSO_4$ (4g/l) | 0.1 |
| Compare the amount of copper in the copper pretreatment step | | | | |
| C5 | Agglomerated | 1 | $CuSO_4$ (2g/l) | 0.1 |
| C6 | as received | 1 | $CuSO_4$ (2g/l) | 0.1 |

Results are presented Figures 2.1a and 2.2a. These Figures are plots of %Au recovered solution versus the cumulative weight of recovered solution for the two comparisons.

Conclusion

Comparison of the form of copper in pretreatment ($Cu^{2+}$ vs $Cu(NH_3)_4{}^{2+}$)

- The best performed columns for Au extraction are those where the ore was:

  (i) pretreated with copper tetrammine (both agglomerated or as received ore; or

(ii) agglomerated and pretreated with $CuSO_4$ .

Comparison of the amount of copper in the copper pretreatment step:

- Halving the copper concentration of the copper-sulfate pretreatment appeared to make little difference to Au extraction rate in the as received ore but reduced extraction rate in the agglomerated ore by about half

**EXAMPLE 3.**

This example relates to leaching of co-agglomerated low-grade oxide ore (~6ppm Au) using columns.

In this example the ore was first pretreated with copper before subsequent thiosulfate treatment was performed. To reduce the number of treatment steps and simplify operation in the field, it may be possible to apply the required copper component by co-agglomerating it (in addition to the cement) in the ore and thus avoid the pretreatment step. Field operation would then require only thiosulfate washing during extraction. To this end a series of co-agglomerated ores were prepared where copper (as copper tetrammine) was added during agglomeration with cement.

Co-agglomeration was performed in the following manner:

Columns 7 & 8 Co-Agglomeration with copper.

To 3 kilograms of ore 18g of cement was added. While this was mixed 400mls of a solution of 0.00156 moles /litre of copper as copper tetrammine was added.

Columns 9 & 14 Co-Agglomeration with copper and ammonium thiosulfate.

To 3 kilograms of ore 18g of cement was added. While this was mixed 200mls of a solution of 0.00312 moles /litre of copper as copper tetrammine was added. In addition to this 200mls of 0.26M ammonium thiosulfate solution was added.

Comparing the extraction efficiency of ores co-agglomerated (besides cement) with either:

(i) small amounts of copper tetrammine (with and without an added copper pretreatment step); or

(ii) a combination of copper tetrammine and thiosulfate.

Leaches were performed in the manner previously described. The following Table (Table 3.1) presents the experimental matrix performed.

Table 3.1

| Column No. | Ore type | Weight (kg) (dry basis) | Ore Bed L/D ratio | Copper Pretreatment Form of $Cu^{2+}$/ concentration equivalent of $CuSO_4$ (g/l) | Thiosulfate Leach Concentration (M) |
|---|---|---|---|---|---|
| C7 | Co-agglomerated with copper tetrammine | 1 | 6.4 | None | 0.1 |
| C8 | Co-agglomerated with copper tetrammine | 1 | 6.6 | $CuSO_4$ (1 g/l) | 0.1 |
| C9 | Co-agglomerated with copper | 1 | 4.9 | None | 0.1 |

| | | | | | |
|---|---|---|---|---|---|
| | tetrammine + thiosulfate | | | | |
| C14 | Co-Agglomerated with copper tetrammine + thiosulfate | 1 | 0.26 | None | 0.1 |
| For comparison | | | | | |
| C1 | Agglomerate | 1 | 6.6 | Copper tetrammine (4g/l) | 0.1 |
| C3 | Agglomerate | 1 | 7 | $CuSO_4$ (4g/l) | 0.1 |
| C11 | Agglomerate | 1 | 0.26 | $CuSO_4$ (4g/l) | 0.1 |

Results are presented in Figure 3.1. This Figure is a plot of %Au recovered versus the cumulative weight of recovered solution.

Conclusion

- The best-performed column (wide column) was that where the ore was co-agglomerated with copper tetrammine and thiosulfate.

- Extraction behaviour decayed towards what appeared to be a limit of about 50%. To determine if the adsorbed copper level was a limiting factor, the column was dosed with a treatment of copper ammine before further thiosulfate washing was undertaken.. Although some subsequent increase in Au extraction occurred, it appeared insubstantial and short-lived. This suggested that, at this crush size, the ore might be limited to an extraction level of about 50-60%.

- The treatments, where the ore was co-agglomerated with copper tetrammine alone (narrow columns C7, C8) showed no particular advantage and were abandoned after about 10 wash cycles. Co-agglomeration in wider columns appeared to have the "initial kick" observed in small-

scale experiments.

EXAMPLE 4.

This example relates to leaching of co-agglomerated low-grade oxide ore (~ 6ppm Au) using columns without using free ammonia.

The inclusion of ammonia or ammonium into the leaching system has a beneficial effect during the early stages of the process of the present invention. However, in some environments the use of ammonium thiosulfate may not be feasible because of its unavailability and the use of free ammonia may also be restricted and sodium thiosulfate would be used as a source of thiosulfate. However, if ammonium sulfate (as opposed to thiosulfate) is freely available it represents a source of ammonia/ammonium. On this basis, co-agglomerates were prepared where copper sulfate and ammonium sulfate were co-agglomerated to mimic the behaviour of copper tetrammine.

Co-agglomeration was performed in the following manner:

Column 12

To 2.2 kg ore was added 11gm cement (5gm/kg). While mixing, 400ml of a solution containing 4gm copper sulfate and 16gm of ammonium sulfate was added. (HIGH level)

Column 13

To 2.4 kg ore was added 12gm cement (5gm/kg). While mixing, 400ml of a solution containing 1gm copper sulfate and 8gm of ammonium sulfate was added. (LOW level)

Table 4.1 presents the experimental matrix performed.

Table 4.1

| Column No. | Ore type | Weight (kg) (dry basis) | Ore Bed L/D ratio | Copper Pretreatment Form of $Cu^{2+}$/ concentration equivalent of $CuSO_4$ (g/l) | Thiosulfate Leach Concentration (M) |
|---|---|---|---|---|---|
| C12 | Co-agglomerated with $CuSO_4$ and $(NH_4)_2SO_4$ HIGH level | 1 | 1.1 | None | 0.1 |
| C13 | Co-agglomerated with $CuSO_4$ and $(NH_4)_2SO_4$ LOW level | 1 | 1.1 | None | 0.1 |
| | | | | | |
| | | | | | |
| For comparison | | | | | |
| C14 | Agglomerate | 1 | 0.26 | Tetrammine (4g/l) | 0.1 |
| C11 | Agglomerate | 1 | 0.26 | $CuSO_4$ (4g/l) | 0.1 |
| C1 | Agglomerate | 1 | 6.6 | Tetrammine (4g/l) | 0.1 |
| C3 | Agglomerate | 1 | 7 | $CuSO_4$ (4g/l) | 0.1 |

Results are presented in Figure 4.1. This Figure is a plot of %Au recovered versus the cumulative weight of recovered solution.

Conclusion

- With a co-agglomerated ore using high levels of Cu and ammonium sulfate, Au extraction behaviour was similar to that of an ore co-agglomerated with copper tetrammine+thiosulfate

EXAMPLE 5

This example relates to leaching of co-agglomerated low-grade oxide ore (~ 6ppm Au) in columns using a copper tetrammine made from copper sulfate, ammonium sulfate and sodium hydroxide and thiosulfate as sodium thiosulfate.

Co-Agglomerated ores were made up as follows:

| Ore Code : | Total ore wt (kg) | Cement (kg/t) | CuSO4 (anhydrous) (kg/t) | Ammonium sulfate (kg/t) | Adjusted with NaOH to make tetrammine | $Na_2S_2O_3.5H_2O$ (kg/t) |
|---|---|---|---|---|---|---|
| 404 | 3 | 5 | 2 | 8 | Yes | 6.6 |
| 405 | 3 | 5 | 2 | 8 | Yes | 3.3 |

Figure 5.1 presents %Au extracted (based on 6ppm of Au in ore) versus weight or volume of recovered lixiviant per wash. Results for Au from the 404 and 405 are compared with previous best performing columns that had co-agglomerated ore with Cu-tetrammine+thiosulfate co-agglomerated ore with CuSO4 + Ammonium sulfate (high)

Conclusion

- The presence of copper tetrammine (made from either method) and thiosulfate in the co-agglomerated ore improves the initial rate of extraction. Slight differences observed between C14 and X-404/X-405 may be accounted for by differences in the thiosulfate concentration used in the co-agglomeration step.

- Based on the recovered solution analysis, the maximum extraction level was in the order of 50-60%.

- At the end of the trials, residues from the best performing columns were fire assayed for Au and the extraction level calculated. This calculation indicated

an extraction of 64-67%, a similar figure to that determined on the as received ore from a cyanide-roll bottle test (~56%). This suggests that the ore crush size may indeed be a limiting factor.

- To clarify this, a sample of as received ore was ring-milled and then leached (in a high concentration thiosulfate, ammonia containing lixiviant system as per experiment 8). In this case, extraction level rose to ~77% confirming a limit on extraction due to crush size.

Many modifications may be made to the process of the present invention described above without departing from the spirit and scope of the present invention.

EXAMPLE 6

This example relates to leaching sulfide ores.

The copper pretreatment conditions were as follows:

- copper tetrammine concentration (M): 0.025M
- ammonia concentration (M): 0.235 - 0.435M
- Total volume (ml): 250

The thiosulfate was conditions were as follows:

- ammonium thiosulfate concentration (M): 0.1
- volume per wash (ml): 100

Two ore/concentrates were examined: Kanowna Belle (X-136) and KCGM (X-133). The following effects were examined:

(i) premilling (by dry ring-milling for 5 minutes (RM))

(ii) varying the form of $Cu^{2+}$ in the pretreatment step ($Cu^{2+}$ cf $Cu(NH_3)_4{}^{2+}$)

Sequential leaches of pyrite concentrates were performed as described above with the incorporation of various treatments. These treatments included:

(i) leaving exposed to air or soaking in thiosulfate for extend periods;

(ii) increasing the concentration of thiosulfate in the wash solution ; and

(iii) re-dosing ore with copper tetrammine.

Results based on solution analyses are presented in Figure 6.

Conclusion

- The highest Au extraction level was ~50-60% using unmilled Kanowna Belle (X-136).

- Premilling appears to inhibit Au extraction although a greater proportion of copper is adsorbed on the ore (60-70% *cf* 30-40%).

- In all cases Cu adsorbed on the concentrate is readily desorbed.

CLAIMS:

1. A process for leaching precious metals from material containing precious metals, which process includes the steps of:

(i) treating the material by oxidising precious metal in the material into a form that is leachable in a subsequent leaching step; and thereafter as a separate step

(ii) leaching the precious metal with a leach solution containing a thiosulfate-based lixiviant.

2. The process defined in claim 1 wherein the material is in the form of ores and concentrates of the ores.

3. The process defined in claim 2 wherein the ores and concentrates are gold-bearing ores and concentrates.

4. The process defined in any one of the preceding claims wherein treatment step (i) includes forming agglomerates of the precious metal-bearing material and an oxidant.

5. The process defined in claim 4 wherein the agglomerates are formed by contacting the material and a solution containing the oxidant.

6. The process defined in claim 5 includes forming agglomerates of the material, a binder, and the oxidant.

7. The process defined in claim 6 includes forming agglomerates by mixing the material (such as an ore or

concentrate of the ore) and the binder and thereafter contacting the mixture with a solution containing the oxidant.

8. The process defined in any one of claims 4 to 7 includes curing the agglomerates.

9. The process defined in claim 8 includes curing the agglomerates in air for a period of at least 24 hours.

10. The process defined in any one of claims 4 to 9 wherein the treatment step (i) includes forming agglomerates of the precious metal-bearing material and an oxidant and a thiosulfate-based lixiviant.

11. The process defined in any one of claims 1 to 3 wherein treatment step (i) includes forming agglomerates of the precious metal-bearing material (with or without a binder) and thereafter contacting the agglomerates with a solution containing the oxidant.

12. The process defined in claim 11 wherein treatment step (i) includes contacting the agglomerates with a solution containing a thiosulfate-based lixiviant.

13. The process defined in any one of claims 1 to 3 wherein treatment step (i) includes contacting the material (without agglomerating the material first) with a solution containing the oxidant.

14. The process defined in claim 13 wherein treatment step (i) includes contacting the material with a solution containing a thiosulfate-based lixiviant.

15. The process defined in any one of claims 5 to 14 wherein the amount of the solution of the oxidant is between 10 and 20% by weight of the weight of the precious

metal-bearing material.

16. The process defined in claim 15 wherein the amount of the solution of the oxidant is between 12 and 15% by weight of the weight of the precious-metal bearing material.

17. The process defined in any one of the preceding claims includes treating the material with ammonia or an ammonium salt, such as ammonium carbonate, to stabilise the oxidant.

18. The process defined in any one of the preceding claims wherein the oxidant is a soluble source of copper ions.

19. The process defined in claim 18 wherein the oxidant is selected from the group consisting of copper sulfate, copper salt, and ammonium complex of divalent copper.

20. The process defined in any one of the preceding claims wherein the thiosulfate lixiviant is selected from the group consisting of sodium thiosulfate and ammonium thiosulfate.

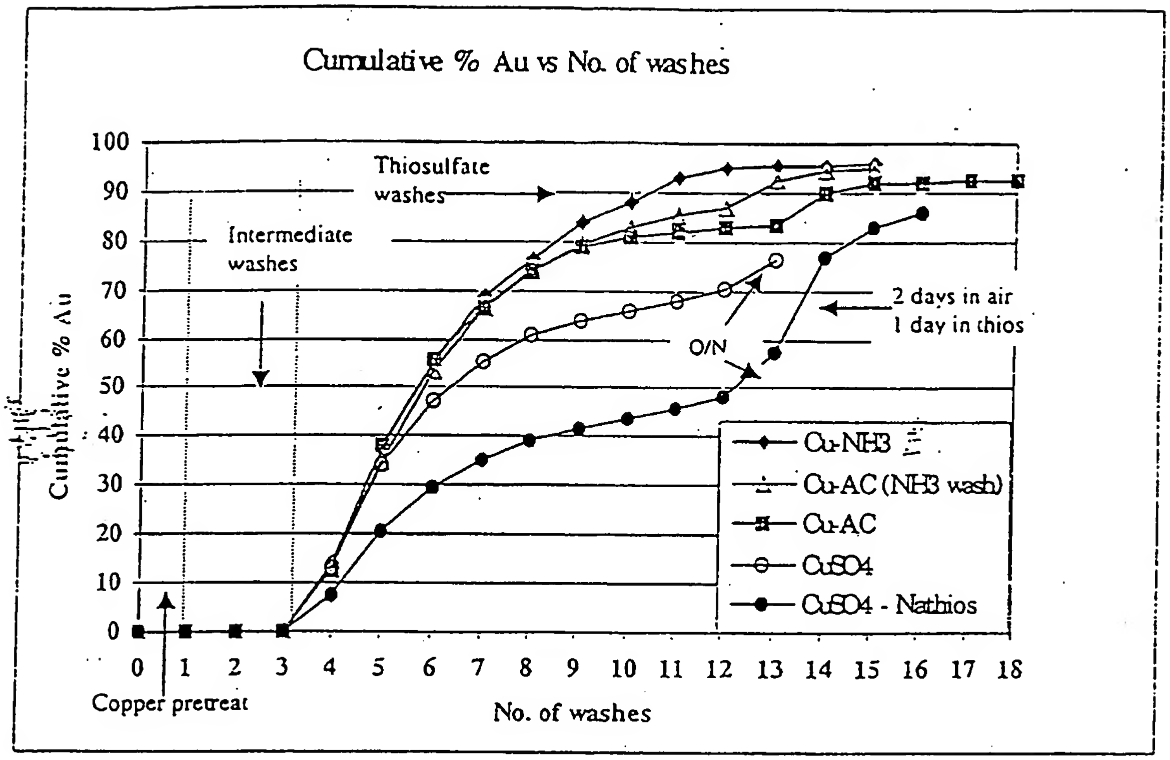
Cumulative % Au vs No. of washes
Cumulative % Au
100
90
80
70
60
50
40
30
20
10
0
0 1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16 17 18
No. of washes
Thiosulfate washes
Intermediate washes
2 days in air
1 day in thios
O/N
Copper pretreat
Cu-NH3
Cu-AC (NH3 wash)
Cu-AC
CuSO4
CuSO4 - Nathios

FIGURE 1.1

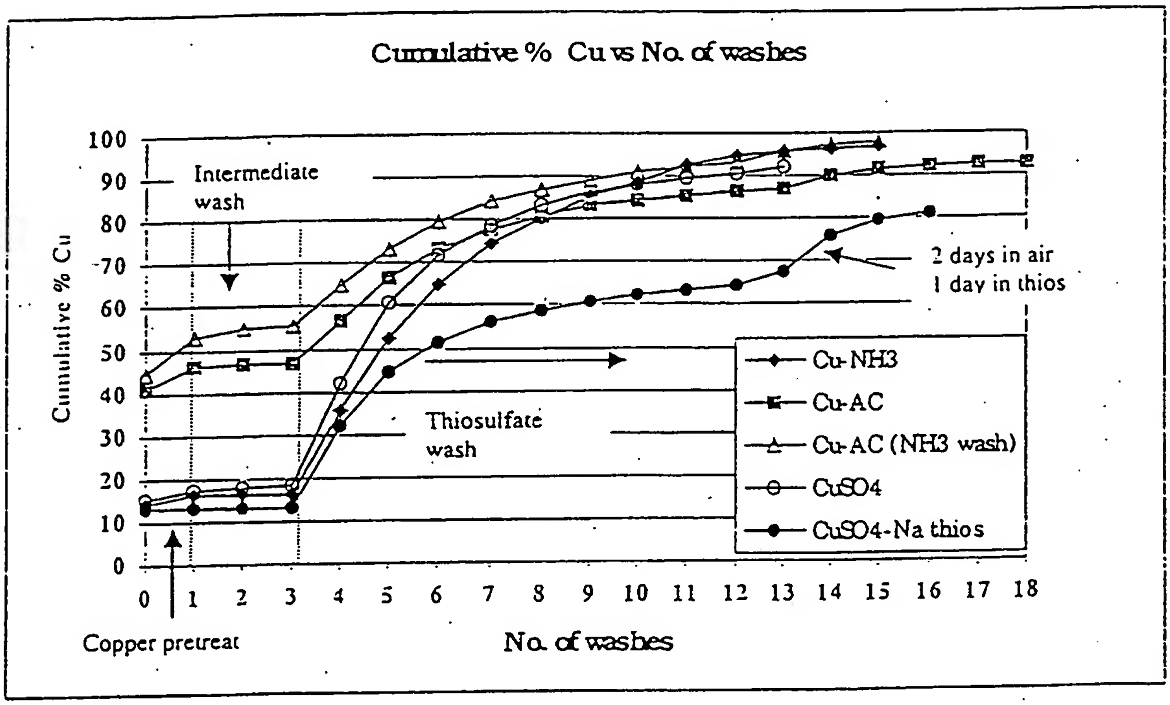
Cumulative % Cu vs No. of washes
Cumulative % Cu
100
90
80
70
60
50
40
30
20
10
0
Intermediate wash
Thiosulfate wash
2 days in air
1 day in thios
Cu-NH3
Cu-AC
Cu-AC (NH3 wash)
CuSO4
CuSO4-Na thios
0 1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16 17 18
Copper pretreat
No. of washes

FIGURE 1.2

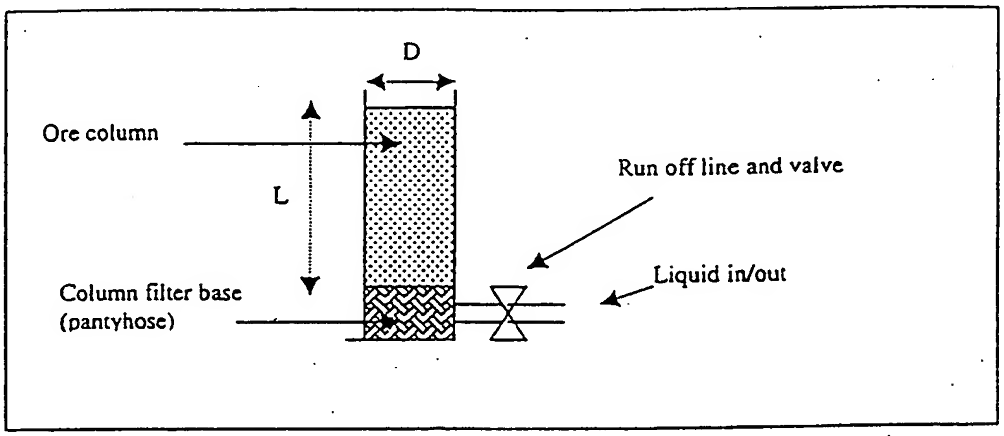
D
Ore column
L
Run off line and valve
Liquid in/out
Column filter base
(pantyhose)

FIGURE 2.1

*Comparison of the form of copper in pretreatment ($Cu^{2+}$ vs $Cu(NH_3)_4^{2+}$)*

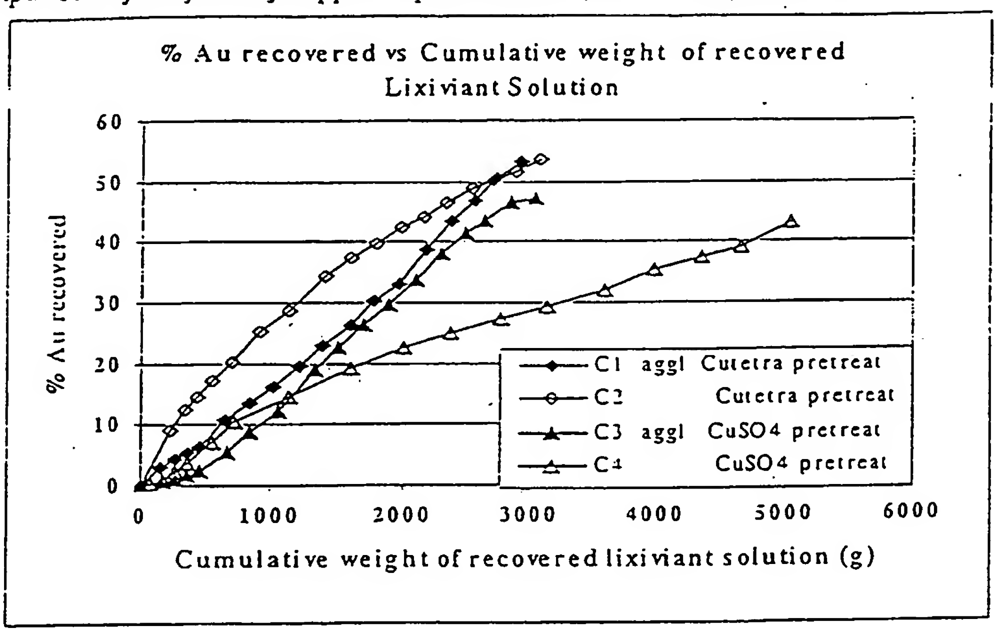


FIGURE 2.1a

*Comparison of the amount of copper (as $CuSO_4$) in the copper pretreatment step (2g vs 4g/l)*

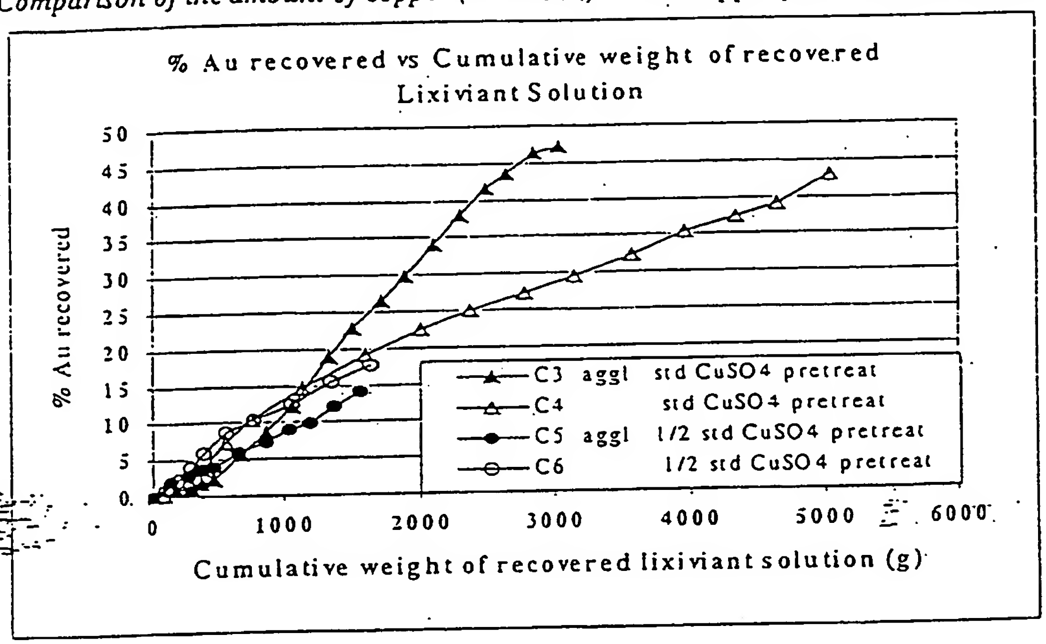


FIGURE 2.2a

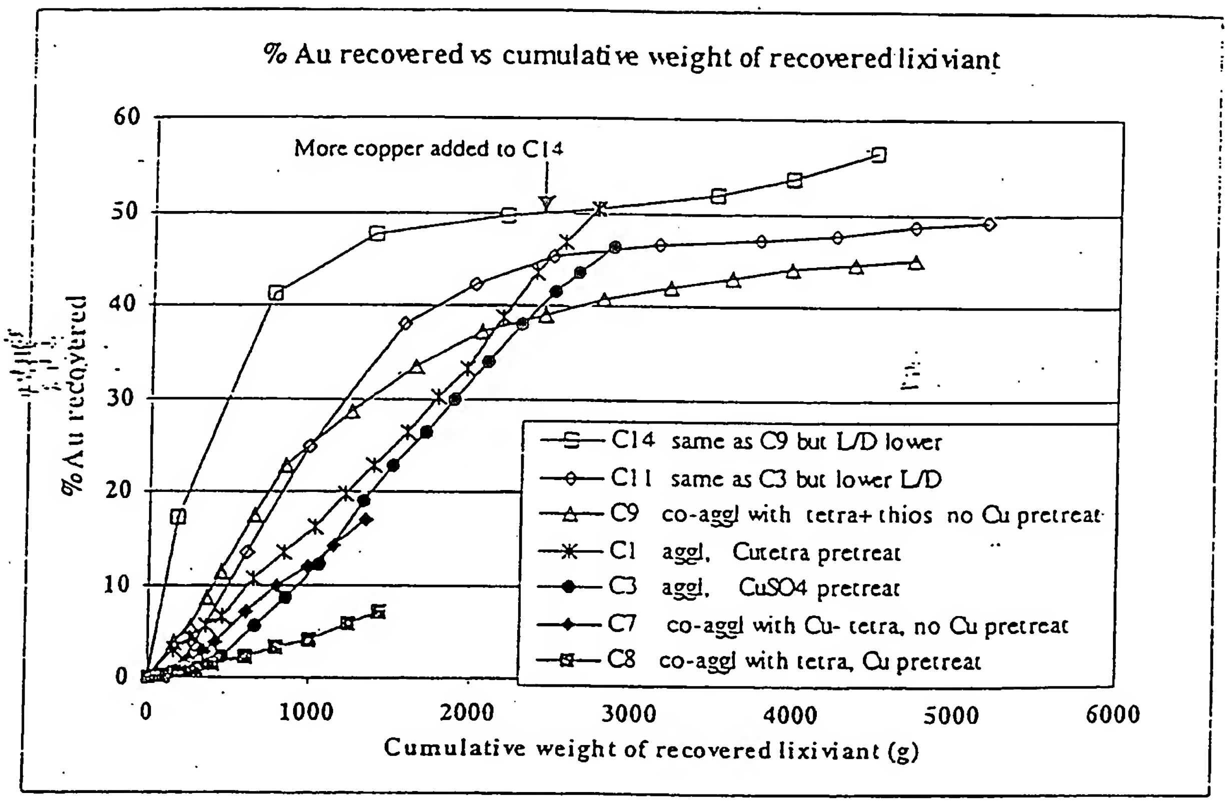
% Au recovered vs cumulative weight of recovered lixiviant
More copper added to C14
60
50
40
30
20
10
0
% Au recovered
0
1000
2000
3000
4000
5000
6000
Cumulative weight of recovered lixiviant (g)
C14 same as C9 but L/D lower
C11 same as C3 but lower L/D
C9 co-aggl with tetra+ thios no Cu pretreat
C1 aggl. Cutetra pretreat
C3 aggl. CuSO4 pretreat
C7 co-aggl with Cu- tetra, no Cu pretreat
C8 co-aggl with tetra, Cu pretreat

FIGURE 3.1

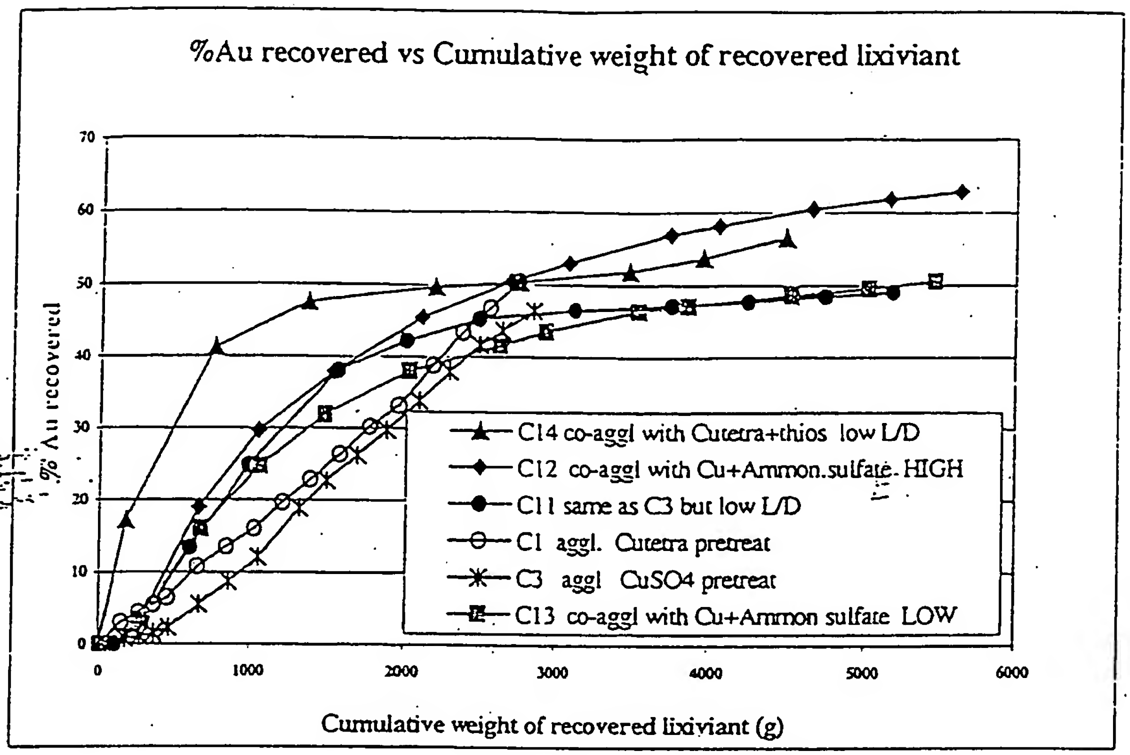
%Au recovered vs Cumulative weight of recovered lixiviant
% Au recovered
Cumulative weight of recovered lixiviant (g)
0
10
20
30
40
50
60
70
0
1000
2000
3000
4000
5000
6000
C14 co-aggl with Cutetra+thios low L/D
C12 co-aggl with Cu+Ammon.sulfate HIGH
C11 same as C3 but low L/D
C1 aggl. Cutetra pretreat
C3 aggl CuSO4 pretreat
C13 co-aggl with Cu+Ammon sulfate LOW

FIGURE 4.1

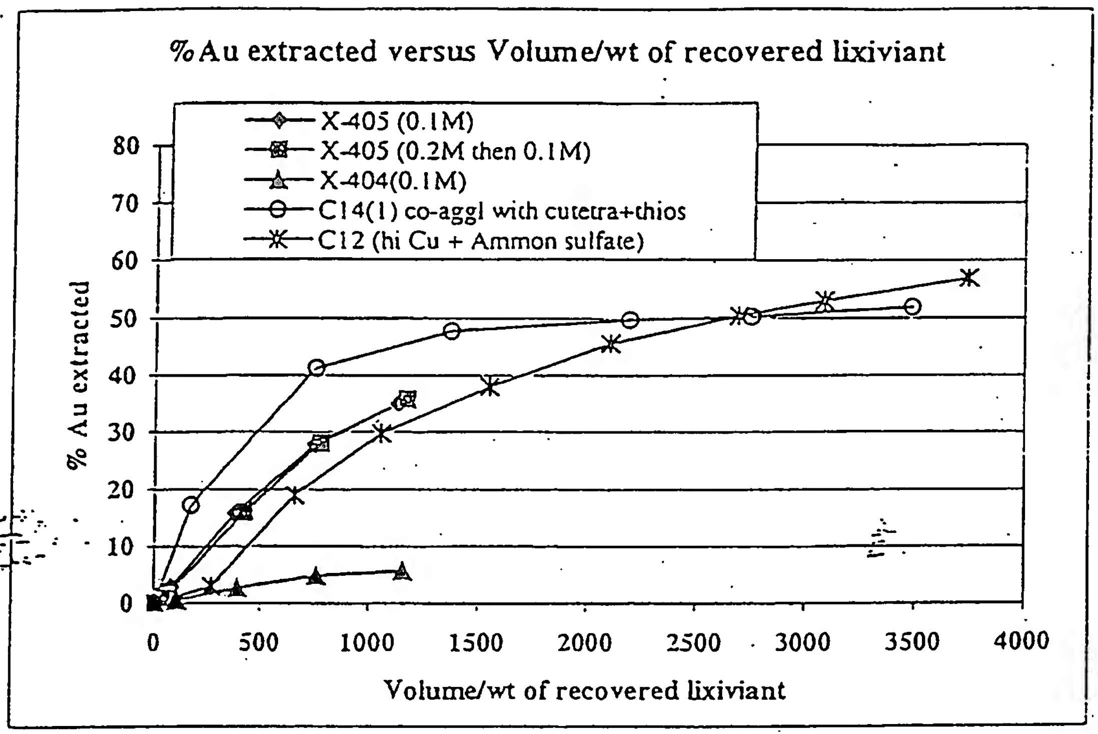
%Au extracted versus Volume/wt of recovered lixiviant
X-405 (0.1M)
X-405 (0.2M then 0.1M)
X-404(0.1M)
C14(1) co-aggl with cutetra+thios
C12 (hi Cu + Ammon sulfate)
% Au extracted
0
10
20
30
40
50
60
70
80
0
500
1000
1500
2000
2500
3000
3500
4000
Volume/wt of recovered lixiviant

FIGURE 5.1

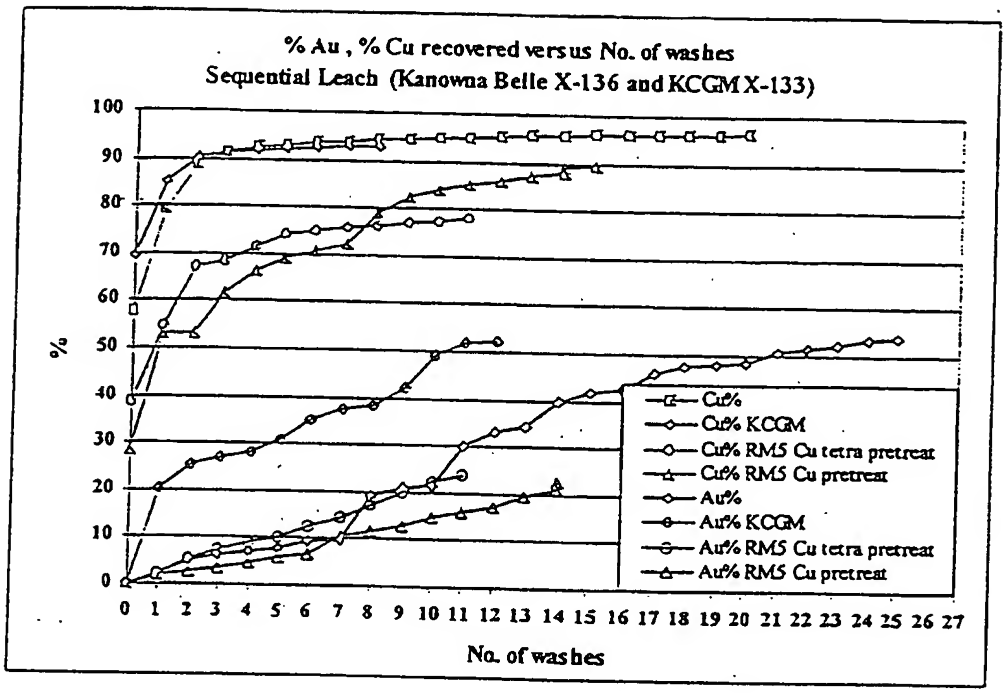
% Au , % Cu recovered versus No. of washes
Sequential Leach (Kanowna Belle X-136 and KCGM X-133)
100
90
80
70
60
50
40
30
20
10
0
%
0 1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16 17 18 19 20 21 22 23 24 25 26 27
No. of washes
Cu%
Cu% KCGM
Cu% RM5 Cu tetra pretreat
Cu% RM5 Cu pretreat
Au%
Au% KCGM
Au% RM5 Cu tetra pretreat
Au% RM5 Cu pretreat

FIGURE 6

**INTERNATIONAL SEARCH REPORT**

International application No.
PCT/AU00/01529

| A. | CLASSIFICATION OF SUBJECT MATTER |
|---|---|
| Int. Cl. [7]: | C22B 3/04, 11/00 |

According to International Patent Classification (IPC) or to both national classification and IPC

**B. FIELDS SEARCHED**

Minimum documentation searched (classification system followed by classification symbols)
IPC C22B 3/00, 3/04, 3/06, 3/12, 11/00, 11/04

Documentation searched other than minimum documentation to the extent that such documents are included in the fields searched
IPC AS ABOVE

Electronic data base consulted during the international search (name of data base and, where practicable, search terms used)
Derwent WPAT

**C. DOCUMENTS CONSIDERED TO BE RELEVANT**

| Category* | Citation of document, with indication, where appropriate, of the relevant passages | Relevant to claim No. |
|---|---|---|
| X | GB 2180829 A (AUROTECH N.L.) 8 April 1987<br>See abstract, page 1 line 64 | 1-4, 11-13, 20 |
| X | EP 522978 A1 (NEWMONT MINING CORPORATION) 13 January 1993<br>See abstract | 1-9, 11, 12, 20 |
| D,X | US 5785736 A (THOMAS ETAL) 28 July 1998<br>See abstract | 1-3, 13, 20 |

[X] Further documents are listed in the continuation of Box C [X] See patent family annex

* Special categories of cited documents:

"A" document defining the general state of the art which is not considered to be of particular relevance
"E" earlier application or patent but published on or after the international filing date
"L" document which may throw doubts on priority claim(s) or which is cited to establish the publication date of another citation or other special reason (as specified)
"O" document referring to an oral disclosure, use, exhibition or other means
"P" document published prior to the international filing date but later than the priority date claimed
"T" later document published after the international filing date or priority date and not in conflict with the application but cited to understand the principle or theory underlying the invention
"X" document of particular relevance; the claimed invention cannot be considered novel or cannot be considered to involve an inventive step when the document is taken alone
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"&" document member of the same patent family

| Date of the actual completion of the international search | Date of mailing of the international search report |
|---|---|
| 5 March 2001 | 7 March 2001 |
| Name and mailing address of the ISA/AU<br>AUSTRALIAN PATENT OFFICE<br>PO BOX 200, WODEN ACT 2606, AUSTRALIA<br>E-mail address: pct@ipaustralia.gov.au<br>Facsimile No. (02) 6285 3929 | Authorized officer<br>MATTHEW FRANCIS<br>Telephone No : (02) 6283 2424 |

| C (Continuation). DOCUMENTS CONSIDERED TO BE RELEVANT | | |
|---|---|---|
| Category* | Citation of document, with indication, where appropriate, of the relevant passages | Relevant to claim No. |
| D,X | US 5354359 A (WAN ET AL) 11 October 1994<br>See column 3 line 61 to column 4 line 11, column 5 lines 8 to 25 | 1-3, 13, 20 |
| X | Derwent Abstract Accession No. 86-189901/30, AU 45769/85 A (PM MINERAL LEACHING) 5 June 1986 | 1-3, 13, 20 |
| X | Derwent Abstract Accession No. 86-194217/30, JP 1127833 A (IN-SITU INC) 16 June 1986 | 1-3, 13, 20 |
| X | Derwent Abstract Accession No. 08269A/04, ZA 7700-840 A (SHERRITT GORDON MIN) 9 November 1977 | 1-3, 13, 20 |
| X | Derwent Abstract Accession No. 83-751239/35, RO 81261 A (INST EPURAREA APELO) 30 January 1983 | 1, 13, 20 |
| X | Derwent Abstract Accession No. 85-300399/48, JP 0208434 A (NIPPON MINING KK) 21 October 1985 | 1, 13, 20 |
| A | GB 2310424 A (FINCH LIMITED) 27 August 1997 | |
| A | US 4369061 A (KERLEY JR) 18 January 1983 | |
| D,A | US 4269622 A (KERLEY JR) 26 May 1981 | |
| A | Derwent Abstract Accession No. 86-194218/30, JP 1127834 A (IN-SITU INC) 16 June 1986 | |

INTERNATIONAL SEARCH REPORT
Information on patent family members

International application No.
PCT/AU00/01529

This Annex lists the known "A" publication level patent family members relating to the patent documents cited in the above-mentioned international search report. The Australian Patent Office is in no way liable for these particulars which are merely given for the purpose of information.

| Patent Document Cited in Search Report | | Patent Family Member | | | | | |
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| EP | 522978 | AU | 18687/92 | BR | 9202534 | CA | 2073589 |
| | | MX | 9203974 | TR | 26634 | ZA | 9204825 |
| | | ZW | 10692 | US | 5246486 | US | 5332559 |
| | | US | 5834294 | | | | |
| US | 5785736 | CA | 2169140 | US | 5536297 | CA | 2209559 |
| US | 5354359 | AU | 49257/93 | BR | 9307876 | CA | 2167571 |
| | | CN | 1098142 | EP | 712445 | MX | 9305978 |
| | | WO | 9504164 | ZA | 9307288 | ZW | 13193 |
| AU | 45769/85 | ZA | 8503701 | | | | |
| JP | 1127833 | NONE | | | | | |
| ZA | 7700840 | NONE | | | | | |
| RO | 81261 | NONE | | | | | |
| JP | 60208434 | NONE | | | | | |
| GB | 2310424 | NONE | | | | | |
| US | 4369061 | CA | 1148748 | US | 4269622 | ZA | 8007196 |
| US | 4269622 | CA | 1148748 | ZA | 8007196 | US | 4369061 |
| JP | 1127834 | NONE | | | | | |

END OF ANNEX